

STABILITY AND SUPPORT IN DEEP MINES

FINAL REPORT

BY

FALCONBRIDGE LIMITED

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Canada Centre for Mineral and Energy Technology,
Energy, Mines and Resources, Canada.

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ABSTRACT

This report is an addendum to the report entitled "Stability and Support in Deep Mines, Progress Report by Falconbridge Ltd., March 1983", and reviews the implementation of a bulk mining method to a shallow dipping orebody at depth.

A section of the Onaping #3 orebody between 3700 Level and 3550 Level was prepared for longitudinal blasthole stoping. The orezone dips at approximately 45° and has a weak norite hangingwall. Primary ground support was by cement grouted cable bolts. To date the first of three 15m cuts has been successfully removed and backfilled.

The study was undertaken to determine the feasibility of the method and if successful to apply it to mining of the Onaping #2 orebody.

Results of this study to date have given insight into the stability of the hanging wall and effectiveness of the hanging wall support design. Modifications to the cable bolt installation procedure, orientation and location for future lifts has resulted from initial results. Blasting design has resulted in acceptable fragmentation.

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1.0 SUMMARY

1.1 INTRODUCTION

In 1979 Falconbridge undertook on a cost shared basis with CANMET the design of an underground bulk mining trial. A section of the Onaping #3 orebody was chosen to be mined by longitudinal blasthole stoping and had progressed to the stage of the first panel slot blast when Onaping Mine was closed on May 21, 1982. The original contract was ammended and a Progress Report was submitted to CANMET (Stability and Support in Deep Mines, Progress Report by Falconbridge Ltd., March 1983). When Onaping was reopened in July 1984, the contract was ammended to allow for completion of the study.

The objective of this report is to fulfill the ammended contract by presenting the data and the subsequent interpretation, acquired through the mining of the first lift of the trial blasthole stope.

1.2 CONCLUSIONS

The principal conclusions from mining of the trial stope may be summarized as follows:

1. The project can be deemed successful as the technical viability of inclined longitudinal blasthole mining has been proven. An inherent difficulty present in this mining method is the instability of the hangingwall. Evaluation of the total impact of the method awaits mining of the second and third lifts.

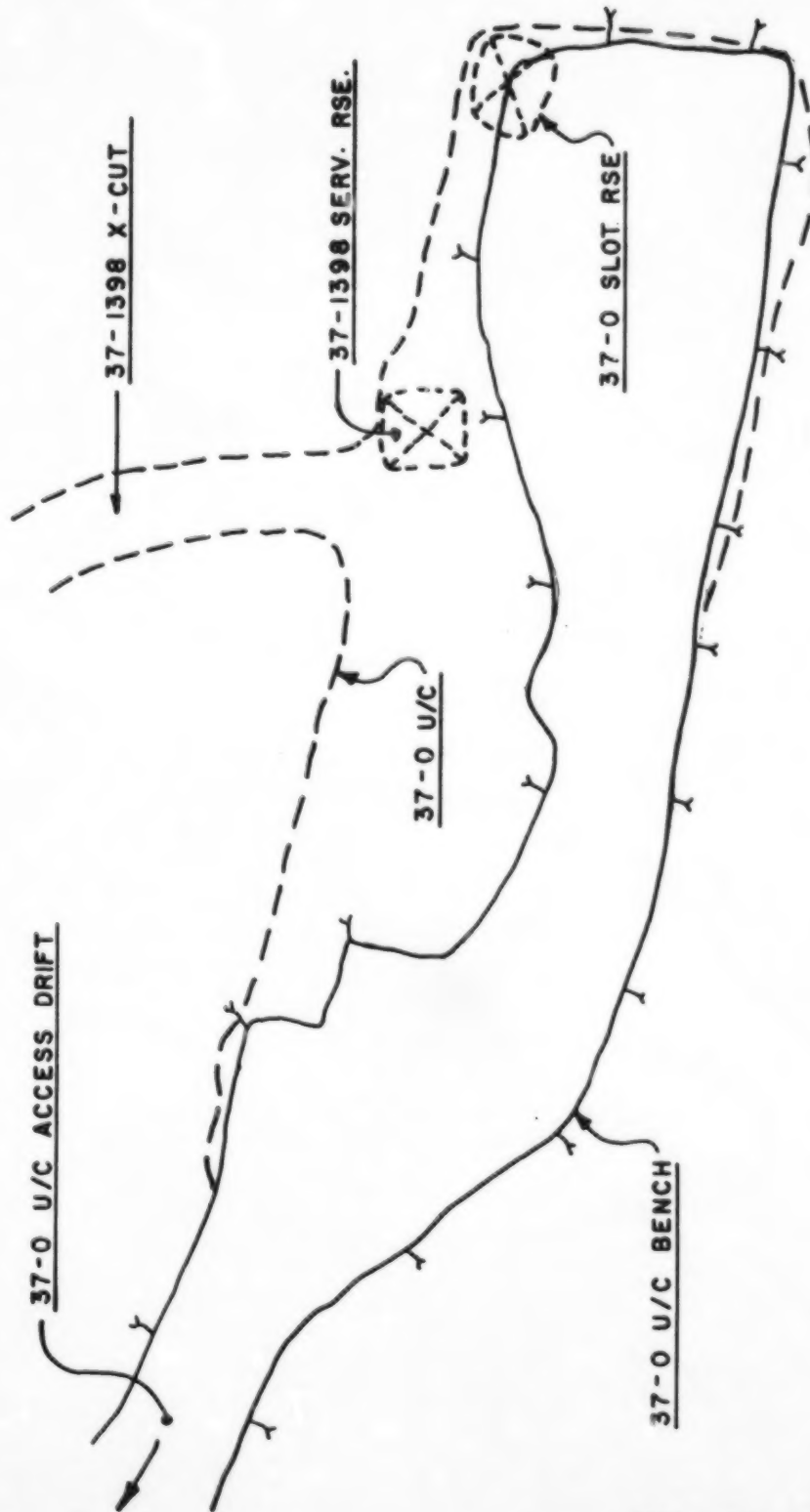
2. The initial design and installation of the hangingwall cable bolt support proved unsuccessful in preventing sloughing of the hangingwall during mining of the first lift. Failure of the support system was attributed to a combination of installation procedure, cable orientation and chemical weathering of the grout. Cable bolt installation for successive lifts has been from undercut/overcut locations and provides support to both undercut back and hangingwall. Hangingwall access drifts have been eliminated, resulting in a savings in development cost.
3. The longhole blast design with inclined boreholes resulted in acceptable fragmentation and smooth hangingwall creation. The change in blasting design from proposed 113mm diameter holes to 51mm holes proved to be successful.
4. Installed instrumentation proved successful in providing data concerning the reaction of the hangingwall to mining. Load cells installed on tensioned cable bolts indicated the nature of load transferral to the bolts and reaction of the cable/grout interface to the increased load.
5. The use of 12:1 cemented fill in the 37+0 undercut to provide a plug for the subsequent pouring of straight tailings fill has proved successful. Monitoring of pressure between the 12:1 fill and fill fence indicated no increase in load during the filling procedure.
6. Numerical modelling gave a qualitative indication of ground reaction that was experienced during mining. Finite element analysis indicated a zone of tension in the hangingwall coinciding with the area that eventually sloughed. An area of theoretical high stress concentration in the footwall experienced working ground when mined.

2.0 DESCRIPTION OF MINING

2.1 CHANGES FROM PLANNED MINING METHOD

The primary mining method remained identical to that outlined in the progress report; that is longitudinal blasthole stoping. However, some changes in method were undertaken and are summarized as follows:

- 1) A three metre bench, approximately six metres wide over the full length of the undercut was taken previous to the mining of the first lift (fig. 1). The bench was filled with waste rock and 32:1 cemented tailings fill with a 0.6 metre thick 8:1 floor.
- 2) Due to the availability of machinery, 51 mm diameter holes were drilled for blasting the orebody rather than the proposed 113 mm diameter holes. Consequently, a tighter drilling pattern was used.
- 3) The footwall row of holes was drilled at approximately 45 degrees. It was expected that ore would not freely flow down this slope and a slusher was used to muck to a location accessible by the LHD unit.
- 4) Due to changes in scheduling and development, waste from the 3550 level waste pass system was not used as backfill for the first lift. However, waste from the 37+2 development was used as backfill.



MPL - GE - 93 D

FIG : 1

PLAN OF 37-0 U/C BENCH

MPL - GE - 93 D

FIG : 1

- 5) In subsequent sublevels no hangingwall drifts were constructed. All cable bolting for successive lifts will be done from the undercuts and overcuts.

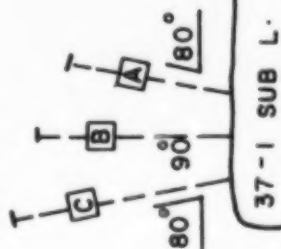
2.2 PRELIMINARY WORK

In October 1984, mining of the stope recommenced with the excavation of a three metre bench in the undercut. The bench was removed under geological control and varied from four to ten metres wide. The bench was backfilled with waste rock and 32:1 cemented tailings fill, 0.6 metres of 8:1 cemented tailings fill was poured as a mucking floor.

At the same time, cable bolts were installed in the back of the first sublevel (fig. 2). Three rows of cement grouted cable bolts were installed over the full length of the undercut. The cables were designed to support the local back and the proposed hangingwall of the second lift. At a later date additional cable bolts to support the hangingwall of the second lift were installed.

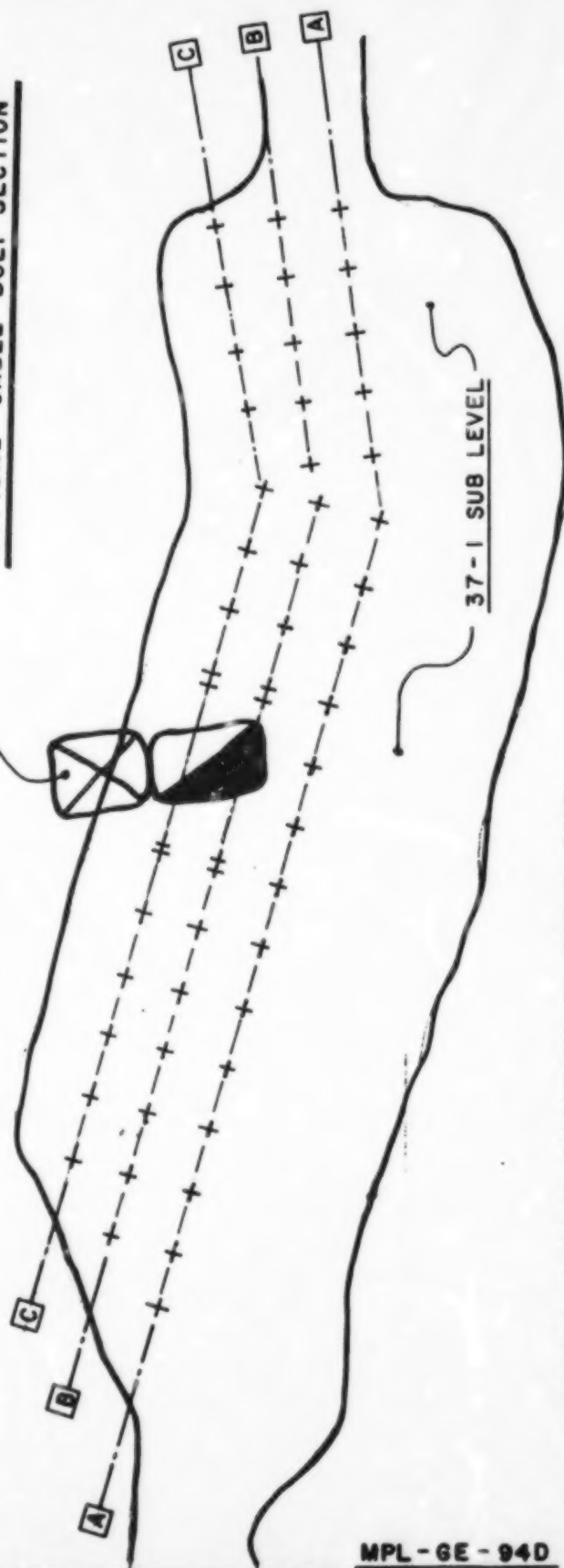
No grout was observed at the collar of the cable bolts installed from the +1 sublevel hangingwall drift. Therefore, supplementary grouting of these cables was performed. Approximately 40 cables were regrouted. However, about one-half of the cable bolts could not be inspected as they had been drilled through the drift floor and covered by broken waste rock.

The benching in the undercut, the installation and regrouting of the cable bolts was completed by late November.



TYPICAL CABLE BOLT SECTION

37-1398 SERV. RSE.



+ = DENOTES COLLAR LOCATION OF CABLE BOLT

MPL-GE-94D

FIG : 2

PLAN AND SECTION OF CABLE BOLTS

IN 37+1 'SUBLEVEL

MPL-GE-94D

FIG : 2

2.3 PRODUCTION

Blasting of the first lift of the stope began in December 1984. Table 1 compares the forecast and actual tonnages of ore from this stope, including the undercut bench and the hangingwall slough. A summary of the blasts is presented in Table 2; the blasts are described in the appropriate sections.

2.3.1 Slot Blast

The slot was removed in two separate blasts, on December 4th and 7th, using 51 mm diameter blastholes with some 113 mm diameter holes drilled prior to the shutdown. The burden and spacing of the 51 mm diameter holes were 1.2 and 1.8 metres respectively. The 51 mm diameter holes were loaded with ANFO and the 113 mm diameter holes were loaded with polystyrene ANFO.

The initial stage of the slot blast removed the hangingwall portion of the slot. Fourteen 51 mm diameter holes and three 113 mm diameter holes were loaded and blasted. The maximum charge weight per delay was 45 kg of ANFO equivalent. A smooth hangingwall resulted from this blast. Fragmentation was acceptable, and negligible secondary blasting was required.

The final stage of the slot blast removed the footwall portion of the slot. Throw from the previous blast was removed from the overcut prior to loading the holes. In addition, some of the holes required redrilling. Approximately thirty 51 mm diameter holes were blasted and the maximum charge weight per delay was 80 kg. No unacceptably large muck was generated but, as with the previous blast, throw occurred into the overcut and required removal before production could commence.

TABLE 1
PRODUCTION SCHEDULE

MONTH	FORCAST (tonnes)	ACTUAL (tonnes)
Oct. 1984*	3000	1794
Nov. 1984*	2400	2401
Dec. 1985	1400	1259
Jan. 1985	3800	4217
Feb. 1985	4300	4193
Mar. 1985	4500	4804
Apr. 1985	4500	4564
May. 1985	<u>3000</u>	<u>3549</u>
TOTAL	26900	26781

* Benching of undercut

TABLE 2
BLAST SUMMARY

BLAST	DATE	kg ANFO	EST.m ³ BROKEN (not incl.slough)	POWDER FACTOR kg ANFO/m ³
H.W. Slot	Dec 4	460	340	1.35
F.W. Slot	Dec 7	700	425	1.65
Lines 29 & 30	Jan 4	425	362	1.17
Lines 27 & 28	Jan 8	425	362	1.17
Lines 26 - 20	Jan 31	1450	1270	1.14
Lines 19 - 10	Mar 22	<u>1875</u>	<u>1722</u>	<u>1.09</u>
TOTAL		5335	4481	1.19

2.3.2 Production Blasts

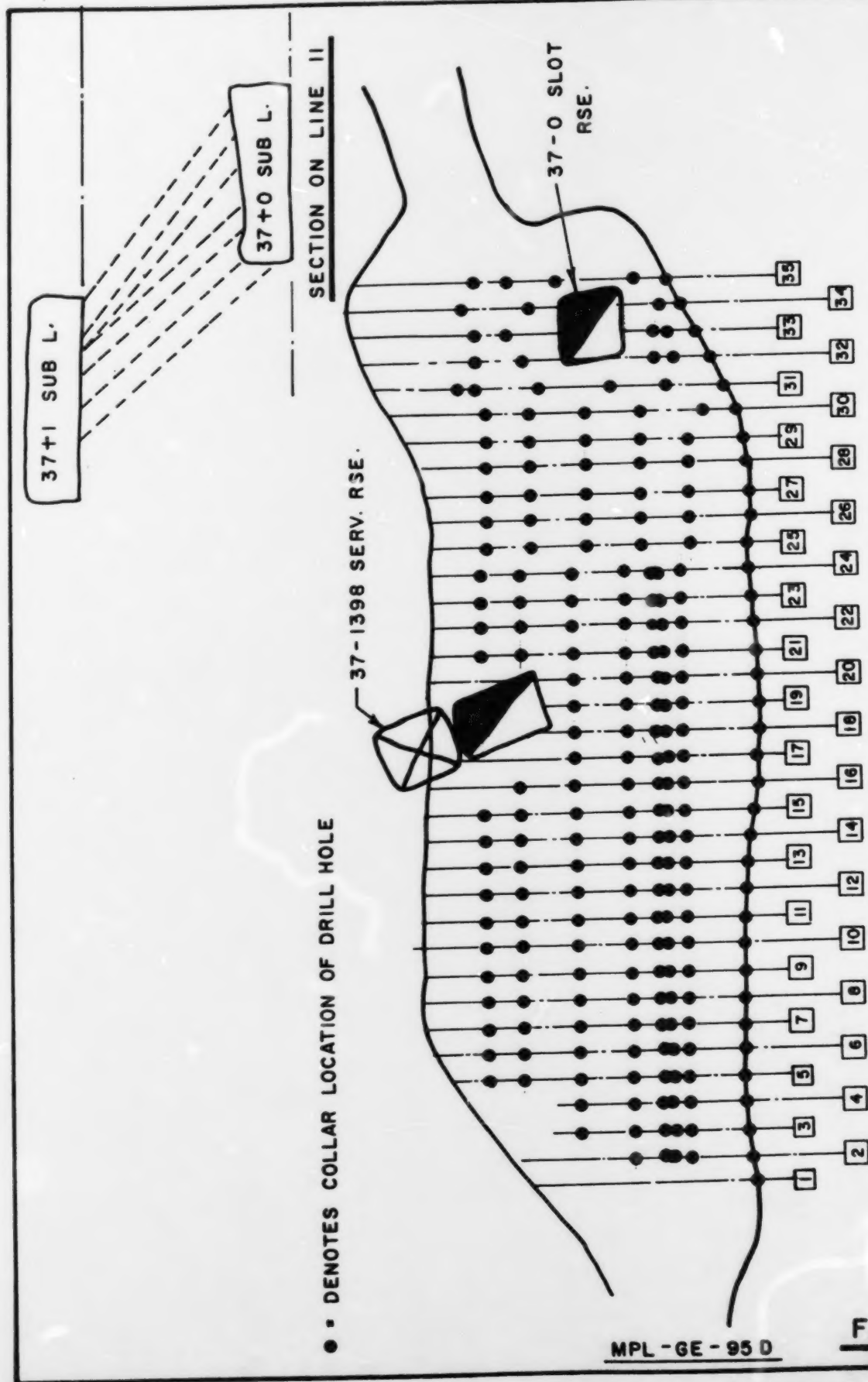
The reduction in drill hole diameter from 113 mm to 51 mm required modification in the drilling pattern. The nominal burden and spacing was changed to 1.2 metres and 1.8 metres respectively, on a parallel drilling pattern (Fig. 3). There was no decking of charges.

The first and second production blasts, on January 4th and 8th, each fired two lines of holes. After the blasts approximately half a metre of cable bolts were exposed in the hangingwall. The finished hangingwall was smooth, and from visual observation and from instrument readings, stable.

Following a seven line production blast on January 31st, 1985 a fall of ground occurred from the hangingwall. The approximate dimensions of the slough were 19 metres along the strike of the overcut, 4 metres into the hangingwall, and 13 metres vertically (Figs. 4, 5 and 6). The approximate volume was 481 cubic metres or, 1540 tonnes.

Instrumentation in the hangingwall drift and in the back of the stope did not indicate any significant movement prior to the fall of ground. It was concluded, from an examination of the structural mapping (Fig. 7) and an inspection of the stope, that the groundfall was structurally controlled by large continuous joints. The groundfall was likely triggered by the blast which allowed the daylighting of the hangingwall structure.

The 1.8 by 1.8 metre pattern of cable bolts at the hangingwall failed to provide adequate support in the groundfall area. Observations concluded that cable slippage had occurred along the steel-cement interface, leaving a trace of the cable



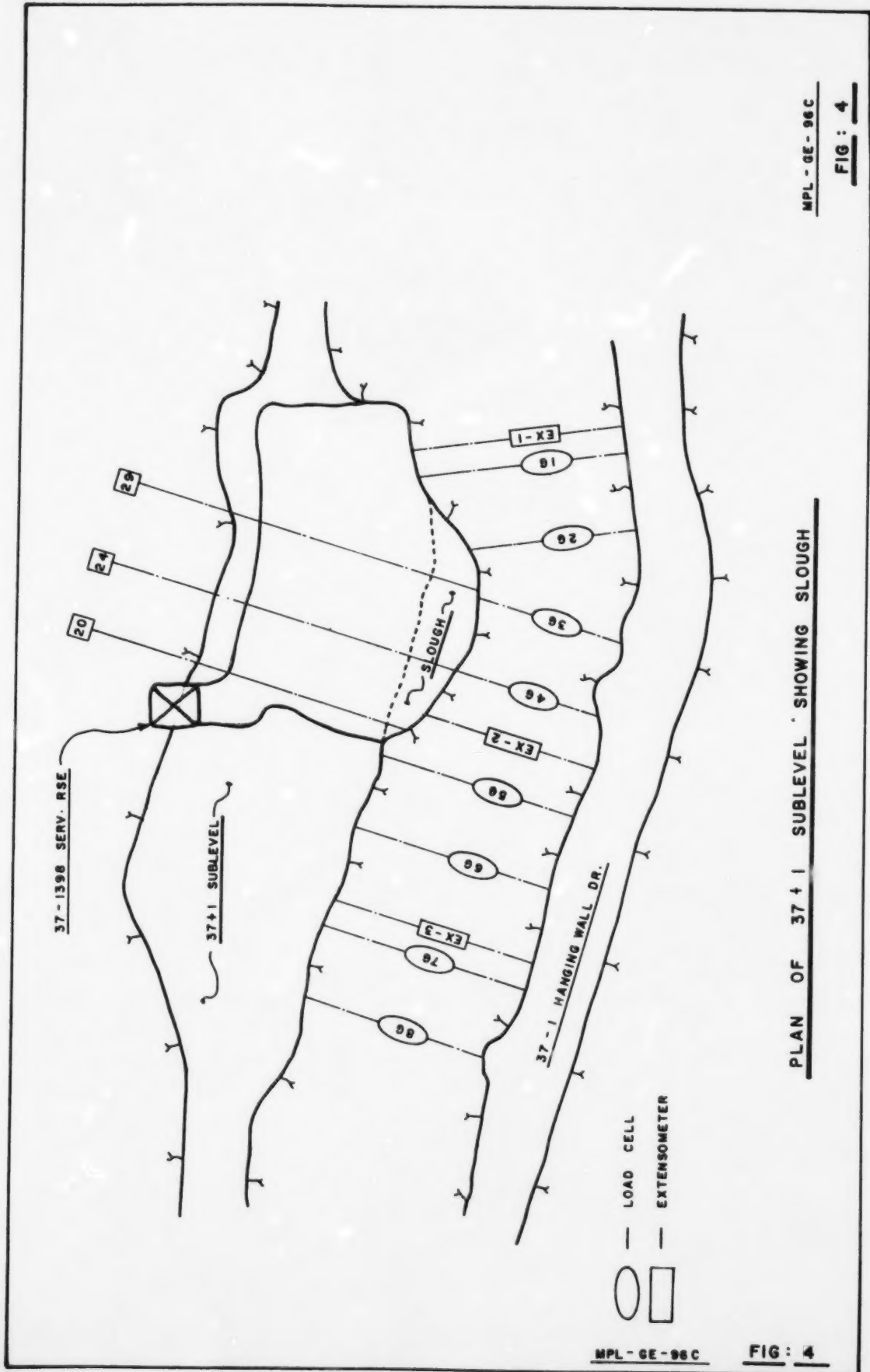
MPL-GE-95 D

FIG : 3

PLAN AND SECTION OF DRILL PATTERN

MPL-GE-95 D

FIG : 3



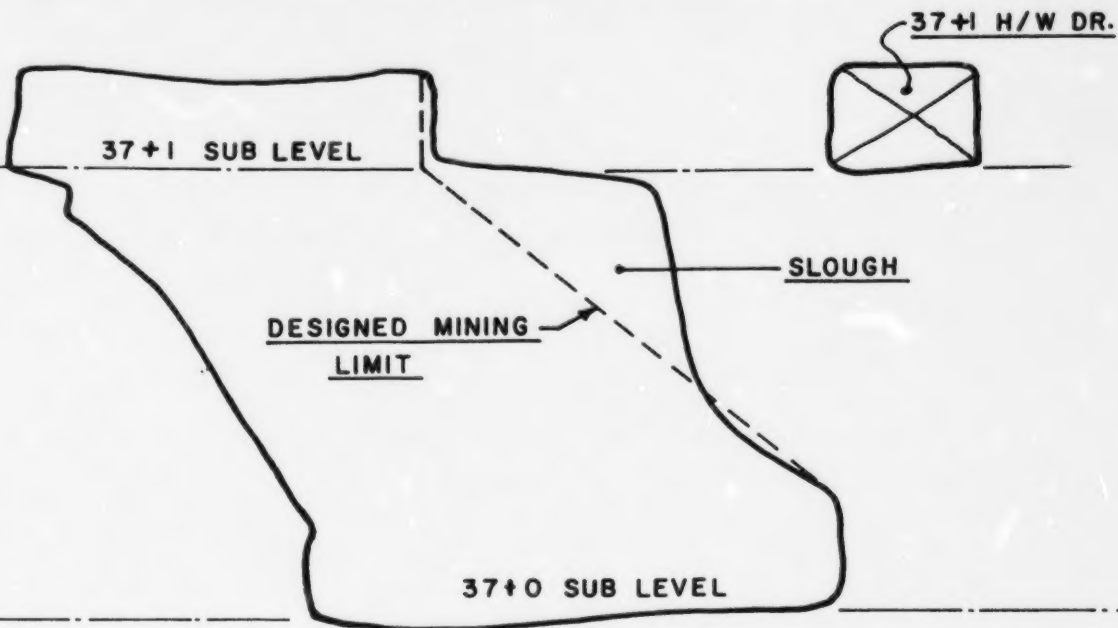
PLAN OF 37+1 SUBLEVEL SHOWING SLOUGH

MPL - GE - 96C

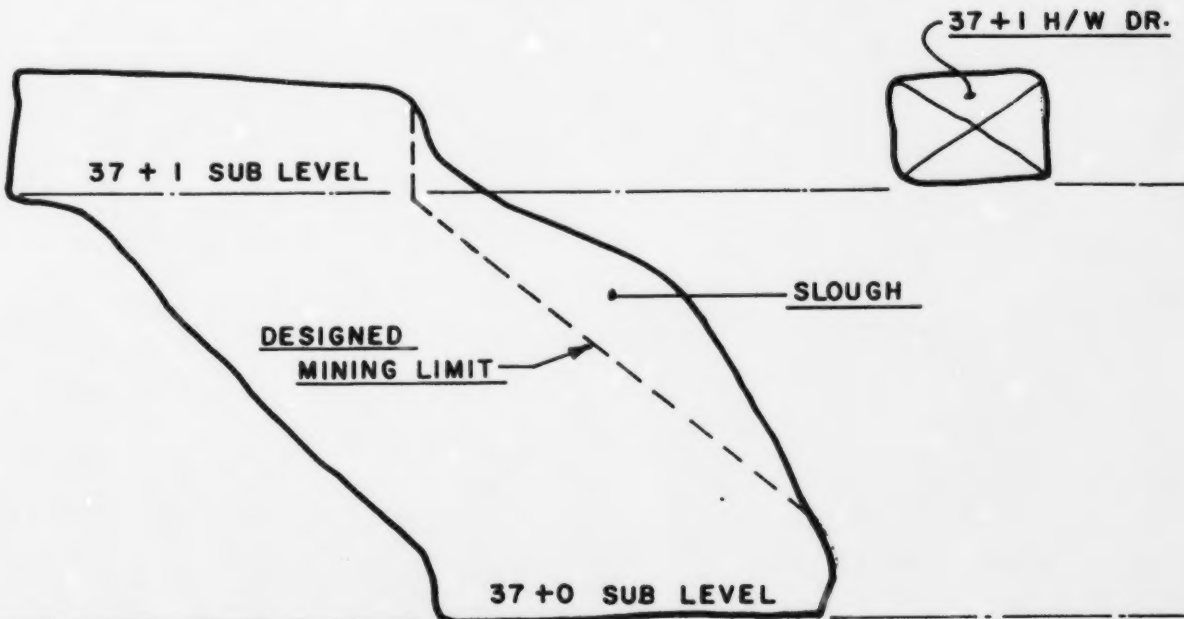
FIG: 4

MPL - GE - 96C

FIG: 4

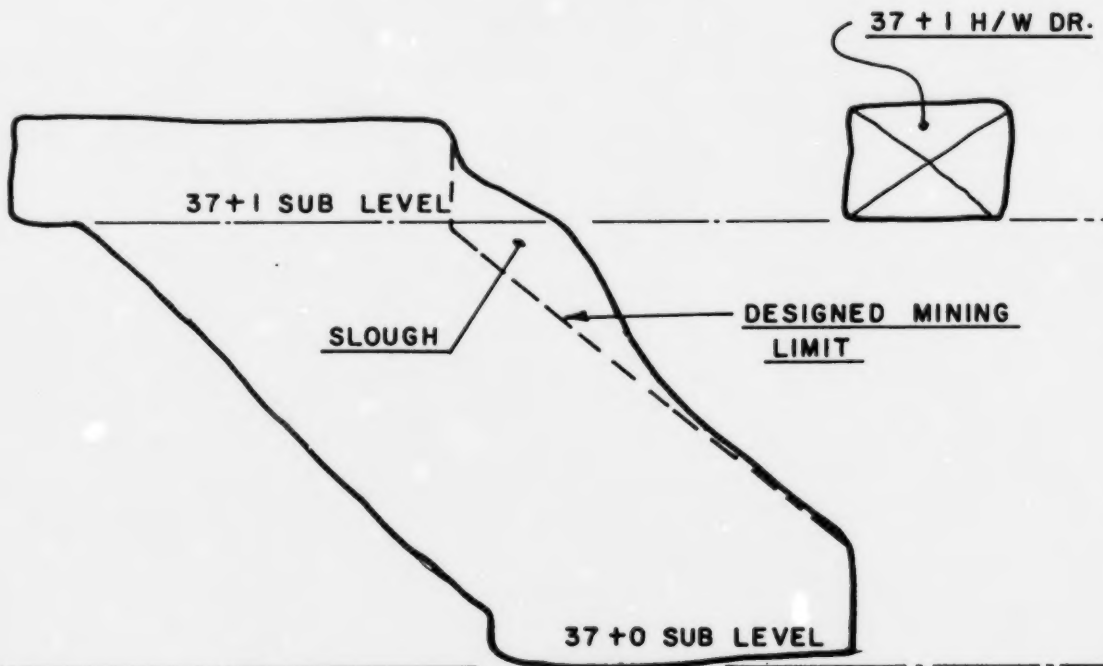


SECTION ON LINE 20



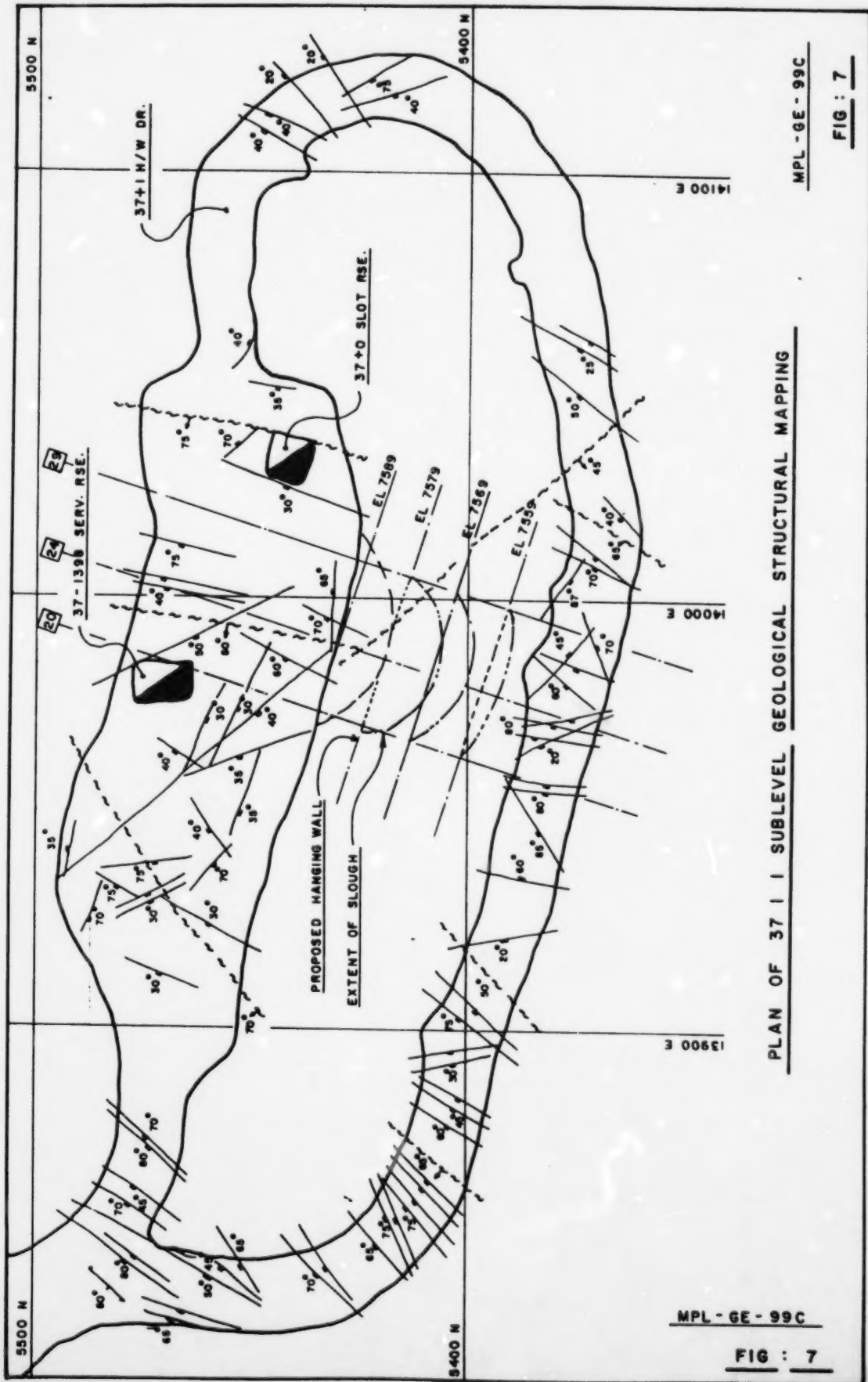
SECTION ON LINE 24

STOPE SECTIONS SHOWING SLOUGH



SECTION ON LINE 29

STOPE SECTION SHOWING SLOUGH



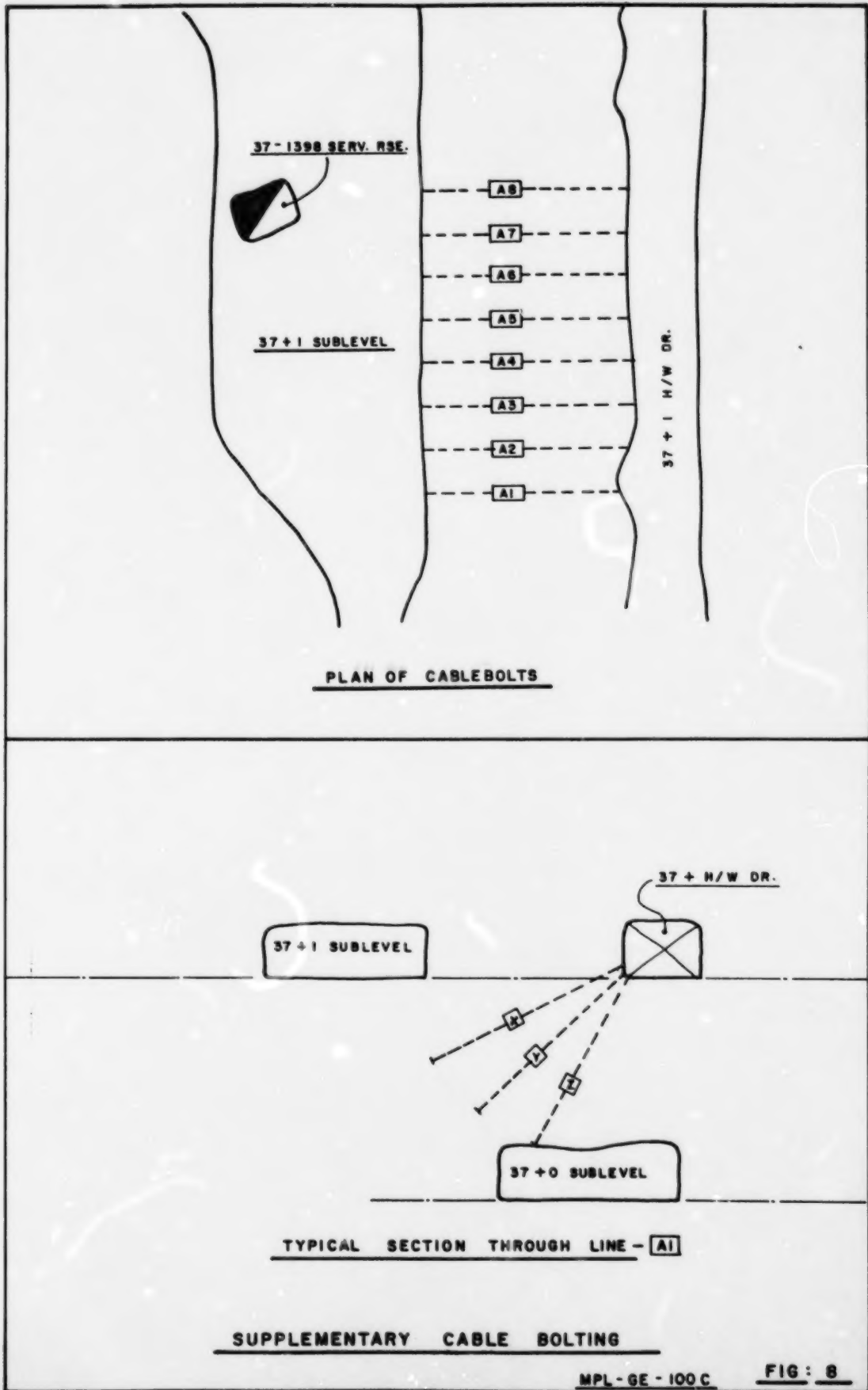
in the remaining cement grout. Cables visible in the hangingwall had not destrandred, which indicates the cables were not heavily loaded prior to the failure.

Instrumentation in the hangingwall drift showed that the hangingwall quickly stabilized after the groundfall. As a remedial measure additional cable bolts were installed from the hangingwall drift into the ore zone (Fig. 8). In addition, timber posts were placed in the undercut to provide local back support and a visual indication of back movement.

Following the cable installation, the grout was allowed to cure for 2-1/2 weeks to ensure adequate strength before blasting. On March 22, 9 1/2 lines were blasted. Again ground sloughed from the hangingwall. The slough appeared to be a continuation of the failure of January 31st, i.e. the hangingwall limit of the slough was now continuous over the full length of the stope. The estimated volume of slough derived from this blast was 368 cubic metres, or 1170 tonnes. As before there was no indication of ground movement prior to the blast, and the hangingwall movement quickly stabilized following the blast.

The behaviour of the original cables and the supplementary cablebolts was identical. That is, neither set appeared to support the ground. Again, the absence of broken cables in the muck pile suggested that the ground slipped off the cables.

Since the remaining stope was not mined due to geological considerations (i.e. an uneconomic grade of ore), further remedial hangingwall support measures were not undertaken. Instrumentation was monitored throughout the mucking and filling processes.



As mucking of the stope continued it was noticed that secondary blasting had loosened the undercut back 4 metres from the brow. A close examination indicated that an approximately 350 tonne wedge could fail (Fig. 9). The wedge was bounded by an infilled fracture which daylighted into the open stope. The wedge was posted and cribbed to provide local support and a visual indicator of progressive movement. On May 12, as the stope was being mucked by remote controlled LHD, the back began to work and small pieces of loose were displaced. Mucking was temporarily halted, the 350 tonne wedge failed and was subsequently removed by the LHD unit.

Approximately 4 metres to the west of the failure an additional fracture was observed to have opened. This fracture was closely monitored but no additional movement was observed.

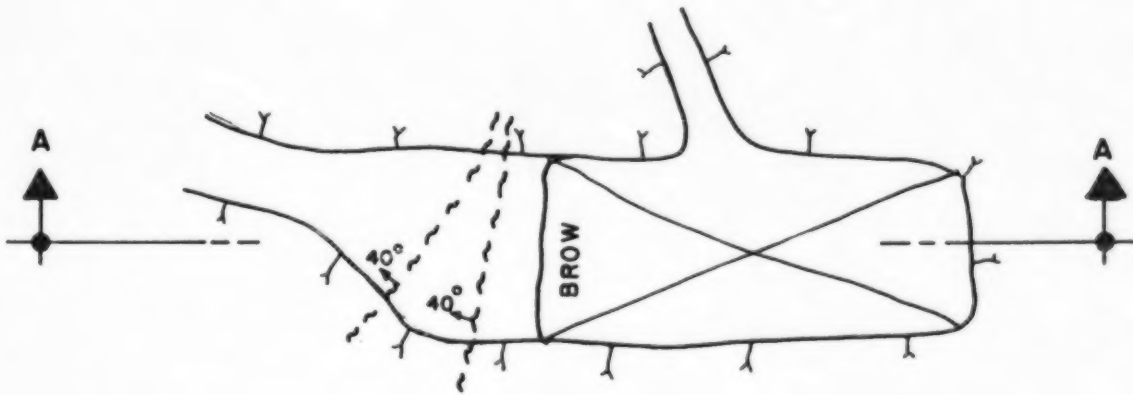
3. MONITORING RESULTS

The graphed results of the hangingwall instrumentation are shown in Figures 10 through 14. The instrument locations are shown in Figure 4. The most significant features are the distinct periods of movement or load registered on the instruments. These periods coincide with the times of major blasts.

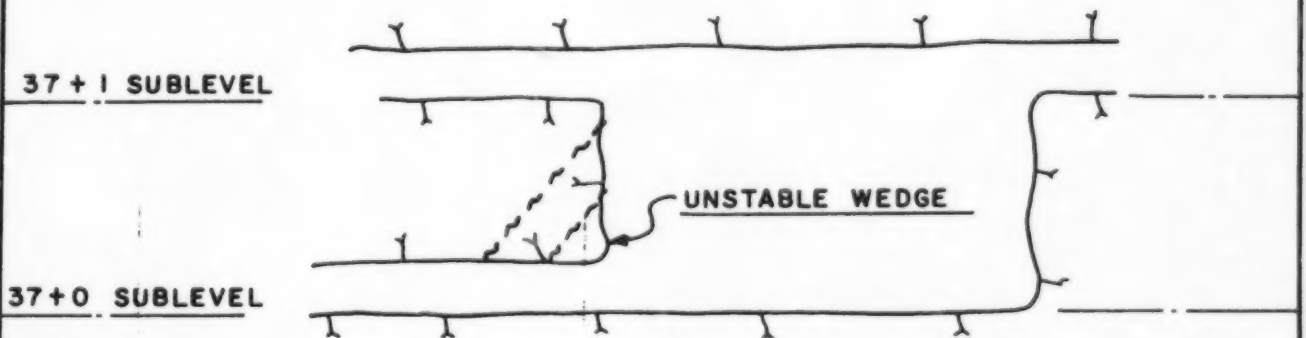
3.1 EXTENSOMETERS

Extensometer 1, the eastern most extensometer, showed an essentially flat response during the period of mining. The extensometer is located in a region where sloughing of the hangingwall did not occur. The response of the extensometer is indicative of the overall stability of the hangingwall.

The centre extensometer, extensometer 2, showed a close correlation between the times of the blasts and registered



PLAN OF 37+0 SUBLEVEL



SECTION A-A

UNSTABLE GROUND IN BROW

—	•	ANCHOR No. 1	30'
- - -	•	ANCHOR No. 2	25'
- - -	•	ANCHOR No. 3	20'
- - -	•	ANCHOR No. 4	10'



GRAPH OF EXTENSOMETER No. 1

MPL-GE-102C

FIG: 10

MPL-GE-102C

FIG: 10

DEPTH

- ANCHOR No. 1 30'
- ANCHOR No. 2 25'
- ANCHOR No. 3 20'
- ANCHOR No. 4 10'

CUMULATIVE MOVEMENT IN INCHES

2.8
2.4
2.2
2.0
1.8
1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2

No. 1

No. 2

No. 4

No. 3

DEC. 1984

JAN. 1985

FEBRUARY

MARCH

APRIL

MAY

GRAPH OF EXTENSOMETER No. 2

MPL - GE - 103C

FIG : II

MPL - GE - 103C

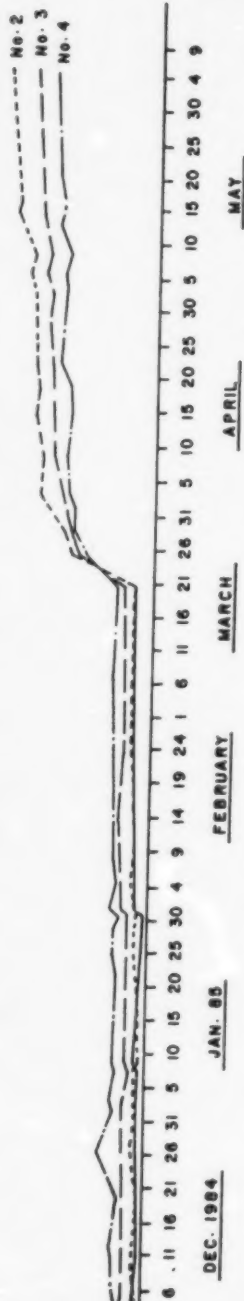
FIG : II

DEPTH

—	ANCHOR No. 1	30'
- - -	ANCHOR No. 2	25'
- - -	ANCHOR No. 3	20'
- - -	ANCHOR No. 4	10'

CUMULATIVE MOVEMENT IN INCHES

1.2
1.0
0.8
0.6
0.4
0.2



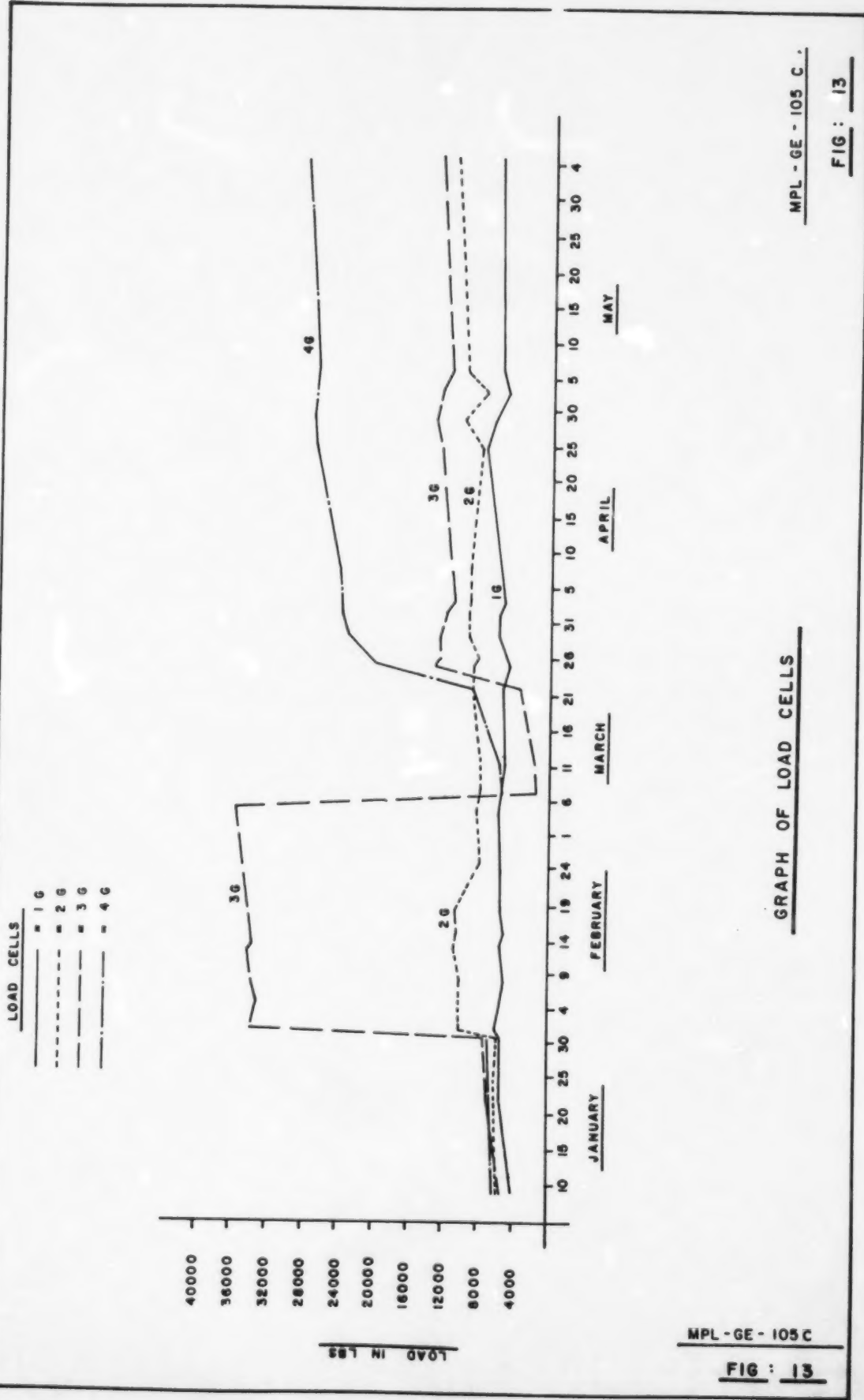
GRAPH OF EXTENSOMETER No. 3

MPL - GE - 104C

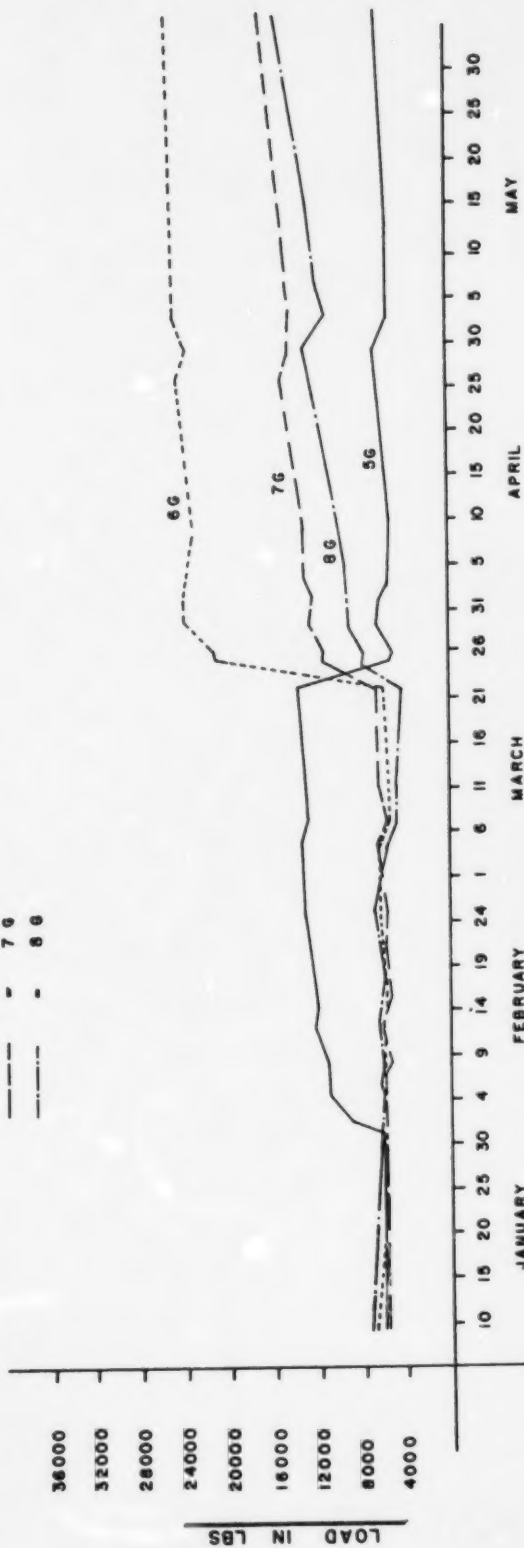
FIG: 12

MPL - GE - 104C

FIG: 12



LOAD CELLS
 - 5 G
 - 6 G
 - 7 G
 - 8 G



GRAPH OF LOAD CELLS

MPL - GE - 106 C

FIG : 14

MPL - GE - 106 C

FIG : 14

movement. The extensometer was unaffected by the slot blasts and first two production blasts, due to the distance between the extensometer and the blast locations. However, the production blast of January 31st caused a significant movement in the hangingwall on extensometer 2. The first anchor, at a depth of 9.1 metres from the collar, was rendered inoperable. This anchor point was likely lost in the initial hangingwall failure. Anchors 2 and 3, at respective depths of 7.6 and 6 metres from the collar, underwent significant movement during the following four days and then moved at a constant rate of 0.015 cm/day until the next blast.

The final production blast triggered the loss of the second anchor. Again anchors 3 and 4 showed a significant movement for a short period of time and stabilized to a small constant rate. It is interesting to note that the movement of anchor 4 did not occur until over a week following the blast.

The western extensometer did not show significant movement until the March 22nd blast. At this time the deepest anchor was lost and the other anchors moved a total distance of 0.7 cm. As with the other extensometers, the movement quickly stabilized to a small constant rate.

3.2. LOAD CELLS

The load cells showed a response similar to that of the extensometers. The January 31st production blast caused an increase in load on cells 2, 3 and 5, and caused the loss of load cell 4. Cell 2, which was to the east of the slough, and cells 6, 7 and 8, which were to the west of the slough, did not register any significant change in load. Only the load on the cells installed near the initial failure were affected.

All of the load cells eventually lost all or part of this acquired load. For cells 2 and 3 the time of the loss of the load is not coincident for the cells, nor does it coincide with a groundfall or with changes in extensometer readings. After relaxing, the cables regained load, although none regained a load equal to the initial level. The behaviour of load cell 5 can be explained by the loss in load having occurred with the final production blast, which may be caused by a decrease in the mass supported by the cable or vibration induced slippage of the cable.

It is suggested that the response of cells 2 and 3 is due to shearing at the cable/grout interface. The cable bolts likely reached a peak loading condition which resulted in shearing at the interface. Sufficient displacement then occurred to decrease the shearing force to a level which could be resisted, which is indicated by the levelling of the curve. As displacement of the hangingwall continued, the load on the cells again began to increase. The ability of the cables to carry a load is indicative of a residual shearing strength at the interface. The ability to carry a load greater than the reduced load level due to blast vibration is indicative that the static shearing strength of the cable/grout interface is greater than the dynamic shearing strength. The ultimate load, therefore, represents the maximum induced force on the cables, and if shearing had again occurred a cyclical loading would be expected.

The March 22nd production blast caused significant changes in load on the cells located near the slough. In general, the cells nearest the slough underwent the greatest change in load. In all cells, except cell 5, the blast caused an increase in load. In cell 5 the registered load decreased dramatically. In view of the behaviour of cells 2 and 3 it is believed that the loss in load is due to shearing at the cable/grout interface.

After the final blast, the load cells showed only a minor increase in load.

4.0 COMPARISON OF MODELLED AND ACTUAL BEHAVIOUR

Since the hangingwall failed it is difficult to derive a meaningful comparison between the monitored loads and displacements and those predicted by numerical modelling. However, some qualitative comparisons can be drawn.

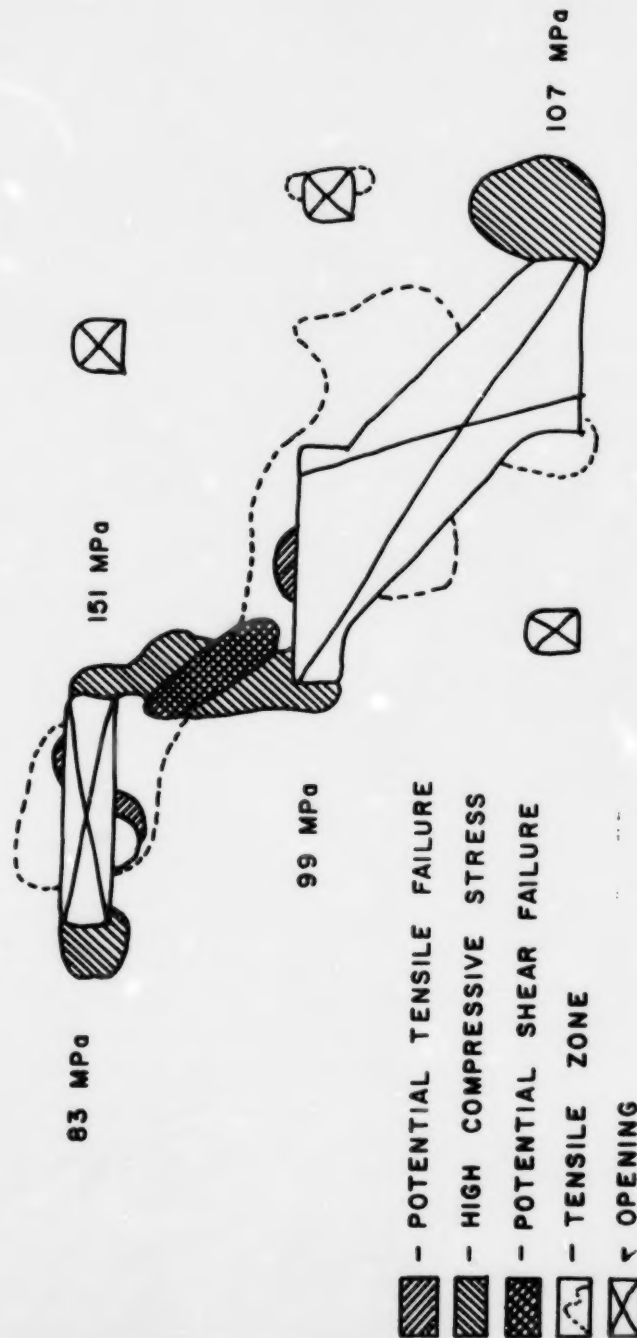
Finite element results showed that after the first lift had been mined, but not backfilled, a tensile zone extended up to 6 metres into the hangingwall (Fig. 15). The general shape of the tensile zone is similar to the extent of the hangingwall failure.

During excavation of the +2 sublevel, working ground was encountered in the footwall side of the stope. Modelling shows this area to be highly stressed but, in addition, shows the hangingwall of the sublevel to be under greater stress. However, there was no evidence of excessive stress concentrations on the hangingwall side of the sublevel.

Comparison of models of the stope at progressive stages of mining awaits further development of the stope. It should be noted that the lack of hangingwall drifts and the presence of a bench in the undercut may influence the behaviour of the ground compared to the geometry analysed by the numerical model.

5.0 CABLE BOLTS

As illustrated by the sloughing of approximately 2710 tonnes from the hangingwall, the designed cable bolt systems did not perform adequately. A 1.8 by 1.8 metre pattern of cable bolts at the proposed mining limit, angled perpendicular to the hangingwall and installed in downholes failed to provide reasonable support to the hangingwall.



TRANSVERSE F. E. MODEL

FIRST PANEL OPEN $K = 1.5$

MPL - GE - 196 D

FIG: 15

MPL - GE - 196 D

FIG: 15

The cable bolt system likely failed to perform due to the following factors:

- (1) geological structures;
- (2) grouting procedures;
- (3) deterioration of grout; and
- (4) differences between the orientations of the cables and the resultant applied force.

As previously discussed, an examination of structural mapping and a physical examination of the stope after the slough indicated that the failure was bounded by distinct structures which daylighted into the opened stope. The structural discontinuity created a large volume of unstable ground.

If the theoretical support capabilities of the cable bolt system is compared to the support requirements of an unsupported slab of ore four metres thick, along the entire length of the stope, the load generated would be sufficient to cause cable failure. However, the mode of failure would be significantly different than that observed. The cables would break at the geological discontinuity, rather than allow the ground to slip off the cables as observed. The observed mode of failure tends to suggest either poor grouting procedures or inherent deficiencies in the designed support method.

Possible poor grouting procedures include:

- (1) insufficient cleaning of holes;
- (2) improper cement content in the grout;
- (3) grease on the cables; and
- (4) insufficient grout in the holes.

Though these factors could impart some influence in the observed failure, the similar behaviour of supplementary cables installed after the initial slough implies that poor grouting procedures do not fully explain the failure mechanism. The installation of the supplementary cables was closely supervised. All of the holes were washed to check for the ability to hold grout, a grout with a water:cement ratio (by weight) of 0.49 was used, the cables were clean, and the holes were grouted from the toe upward. The supplementary cables also allowed the ground to slip from of the cables, therefore it can be concluded that the failure is due to more than poor grouting procedures.

When non-breakthrough downholes are used for cable bolting it is difficult to prepare the holes for grouting. It is difficult to adequately clean the holes of drill cuttings and a few feet of cuttings can be expected to lodge at the toes of the holes. The immediate toe of the hole will be insufficiently grouted and thereby provide little support. However, this does not account for a four metre length of cable to inadequately support the ground.

The initial pattern of cable bolts was installed in 1982, before the closure of Onaping Mine, and therefore the cable bolts were in place approximately three years before mining of the stope commenced. The cable bolt grout is essentially a hardened cement paste composed of Normal Portland cement and is quite susceptible to sulphate attack. Over this time, sulphates in the hangingwall may have attacked the grout causing a deterioration at the borehole annulus and allow the ground to slip off of the cables. However, the similar behaviour of the supplementary and the initial cables precludes the inference of sulphate attack being the sole cause of the failure.

The cable bolt system was designed such that the cables formed a 1.8 by 1.8 metre pattern on the hangingwall with the cables oriented approximately perpendicular to the hangingwall. However, the resultant force, essentially due to gravity, acted in a downward direction. The direction of principle strength of the cable was oriented at an approximately 45 degree angle to the load, therefore debonding at the cable/grout interface may have occurred resulting in the sloughing of the unstable ground.

The failure of the cable bolt system to contain the sloughed ground was not due to a single deficiency in the designed system but rather was due to a combination of the above factors. At this time it is not possible to ascertain the relative importance of these factors. Judgement can be better performed after mining the successive lifts.

6.0 BACKFILLING

The first lift of the stope was backfilled with a combination of 32:1 cemented tailings fill and uncemented tailings fill. Fill fences were installed in the two access drifts on the 37+0 sublevel and 32:1 cemented tailings fills was poured from the 37+1 sublevel to an elevation of 1.5 metres above the undercut brow. Uncemented tailings fill was poured from this elevation to the 37+1 sublevel. A mucking floor on the 37+1 sublevel was constructed from a combination of waste rock and 8:1 cemented tailings fill.

A fill pressure cell was installed on a fill fence. The cell was oriented vertically and the pressure was monitored throughout backfilling. No increase in pressure was observed.

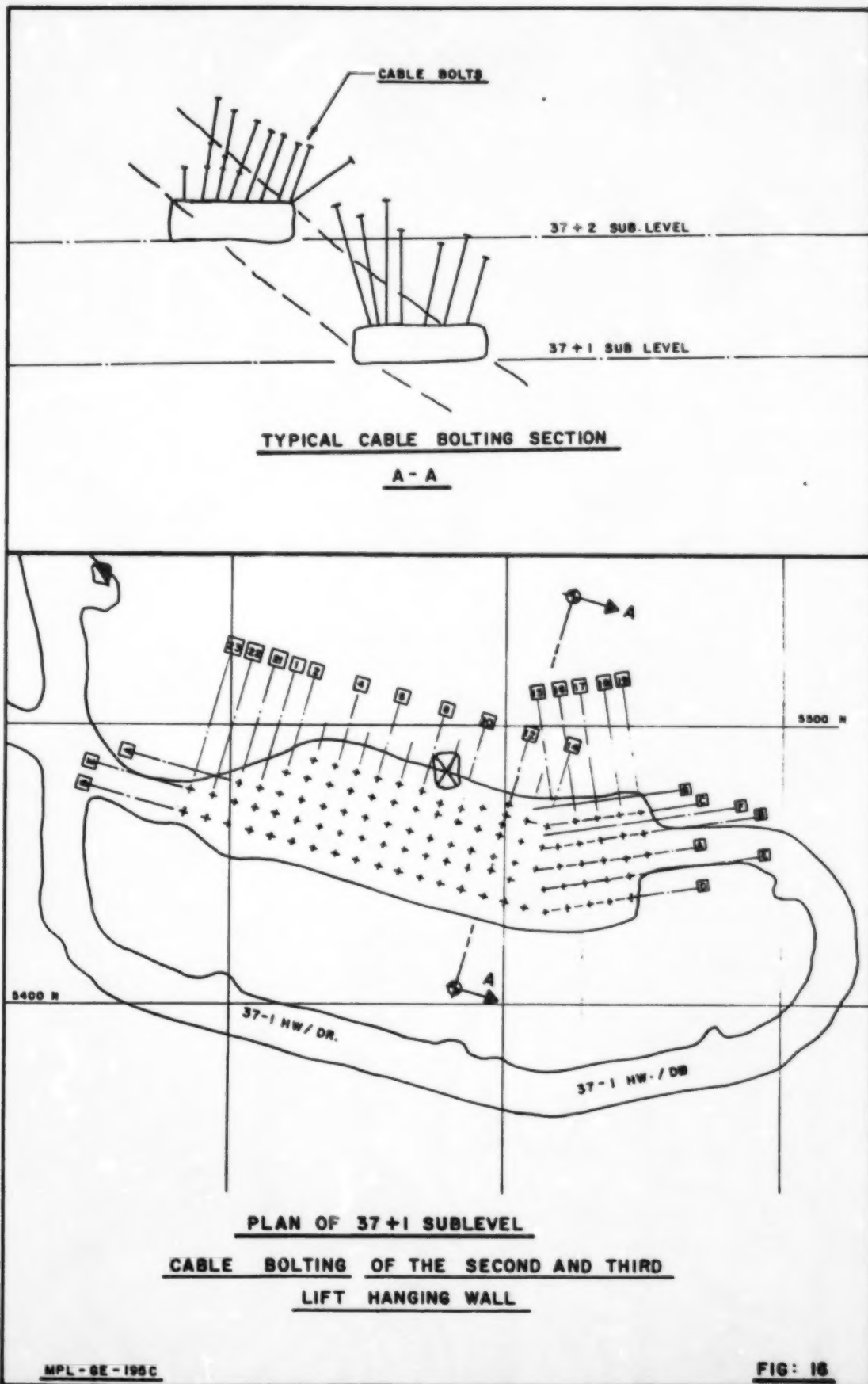
7.0 FUTURE MINING PLANS

Development work for the second lift has begun. A significant change in support design occurred for the successive lifts. Previous to the mining of the first lift it was decided that a hangingwall drift would not be constructed on the +2 and +3 sublevels. It was felt that the possible advantages of cabling from the hangingwall were outweighed by the added expense of constructing such a drift. Therefore, all cabling of the second and third lifts involve installation from the panel undercuts and overcuts (Figure 16).

The hangingwall support of the second lift involved the installation of 7.6 metre cables countersunk to depths appropriate to intersect the proposed mining limit and anchor in the hangingwall.

The cable bolt pattern installed from the +2 sublevel was designed to provide local back support during the mining of the second lift and to provide hangingwall support during mining of the third lift. 7.6 metre cables countersunk into the hangingwall provide support during the mining of the third lift. 4.5 metre cables installed in the collars of the same holes provide local back support for the overcut of the second lift. As opposed to the use of a single long cable in each hole, the split cable design will prevent hang-up of muck when the lift is blasted and provide a smoother hangingwall.

On both sublevels the lines of cable bolts were spaced at 2.4 metre intervals to allow blasthole drilling between the cable bolt lines.



The cable bolt design should eliminate some of the difficulties described with cabling from the hangingwall drift. More importantly, the cable bolts for the second and third lifts are oriented in the direction of the major force acting on the hangingwall. The possibility of poor grouting is reduced as upholes are easier to clean and grout is pumped from the collar upward, ensuring complete grouting of the hangingwall cable bolts.

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